Deep Yellow Limited

ASX Announcement **ASX: DYL**

21 December 2012

SHIYELA IRON PROJECT – SCOPING STUDY UPDATE

KEY POINTS

- **Mintrex Pty Ltd has completed a scoping study for the Shiyela Iron Project, with economically encouraging results.**
- **The capital cost estimate for a mine producing 2 Mtpa of concentrate and at a confidence level of +/-30% was US$367 million whilst operating costs were estimated at US$63.20 per tonne of concentrate FOB.**
- **Plant capital cost is US$207.3 million with the remainder made up of mining-related capex and infrastructure.**
- **Three types of ore are to be processed in the plant, being magnetite, magnetite with significant hematite tails, hematite with minor magnetite tails.**
- **The magnetite can be produced at 68 to 69% Fe with a grind of 80% -250 micron or finer.**
- **The hematite can be produced at 61% Fe with a low gauss WHIMS at 3000 gauss and with a grind of 80% passing 250 micron.**
- **The resource database has been reassessed to incorporate low magnetite – high hematite material (initially considered waste) as potential ore grade material.**
- **The result will allow the company to pursue its previously announced partner strategy for the project.**

Deep Yellow Limited (DYL or the Company) is pleased to announce that Mintrex Pty Ltd (Mintrex) has completed a Scoping Study for its Shiyela Iron Project in Namibia. The Project, which is held by Shiyela Iron (Pty) Ltd (Shiyela Iron), a 95% owned subsidiary of DYL and DYL's Namibian empowerment partner, Oponona Investments (Pty) Ltd (5%), is located entirely within EPL 3496 which is held by DYL's wholly-owned Namibian subsidiary, Reptile Uranium Namibia (Pty) Ltd (RUN) (see Figure 1). (Shiyela Iron was recently provided with a Notice of Preparedness to Grant a Mining Licence (MLA176) by the Ministry of Mines and Energy of the Republic of Namibia.)

The Scoping Study was based on the results obtained from a comprehensive three-stage testwork programme which allowed the development of process flowsheet and preliminary plant design (see Figure 2). The plant design is based on recovering 2 million tonnes per year of product/s from three types of ore in one cost effective design. The overall capital cost is estimated at $366.8 million which is roughly $190/annual tonne of concentrate and estimated operating costs of U$63.02/tonne on a +/-30% level of confidence.

The Fe yield to a dry preconcentrate of both magnetite and hematite is 90% to 95% and with a combined magnetite and hematite Fe yield in the wet plant of 90% gives an overall Fe yield of at least 80 to 85%. The combination of dry MIMS and WHIMS means that the effective yield from the magnetic proportion can be regarded as 100%.

The metallurgical yield from the resource, on the current data, suggests:

- An overall yield of 85% Fe (roughly 100% from DTR magnetic Fe and 70% from hematite Fe).
- That the Fe in the magnetic proportion measured by Davis Tube be assumed to be 100% recovered at a grade of 68% Fe – though on occasions the grind may need to be lowered from 250 micron to 125 micron to reach grade.
- That the remaining Fe be assigned to a hematite stream at 61% Fe. This may need two stages of concentration – rougher and cleaning.
- The final product grade is a mixture of magnetite and hematite in the proportion determined by the head Fe in the ore body. However, the plant is designed so that the magnetite product and hematite product are produced separately and can be mixed as desired.

Greg Cochran, DYL's Managing Director commented that the results of the Scoping Study met the Company's expectations and demonstrated the Project competitive cost position. "We are pleased with the outcome of the testwork and the study which, together with the recent offer of a Mining Licence from Namibia's Ministry of Mines and Energy, places the project in a very strong position. We have also been conservative in our approach and I believe there is still room for improvement in capital and operating costs, as well as the potential to further enhance the resource base."

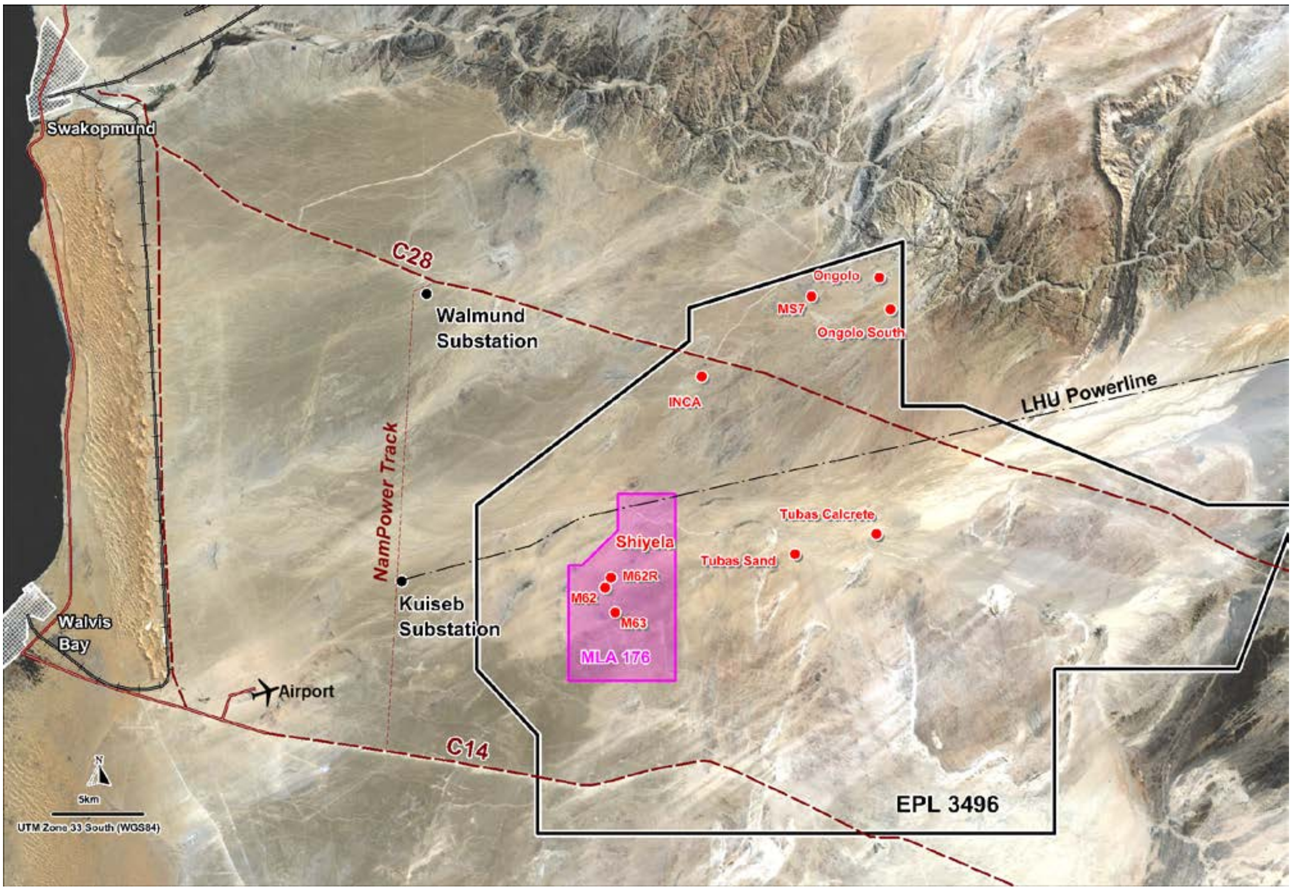

Figure 1: Shiyela Iron Project – MLA and Local and Infrastructure Plan

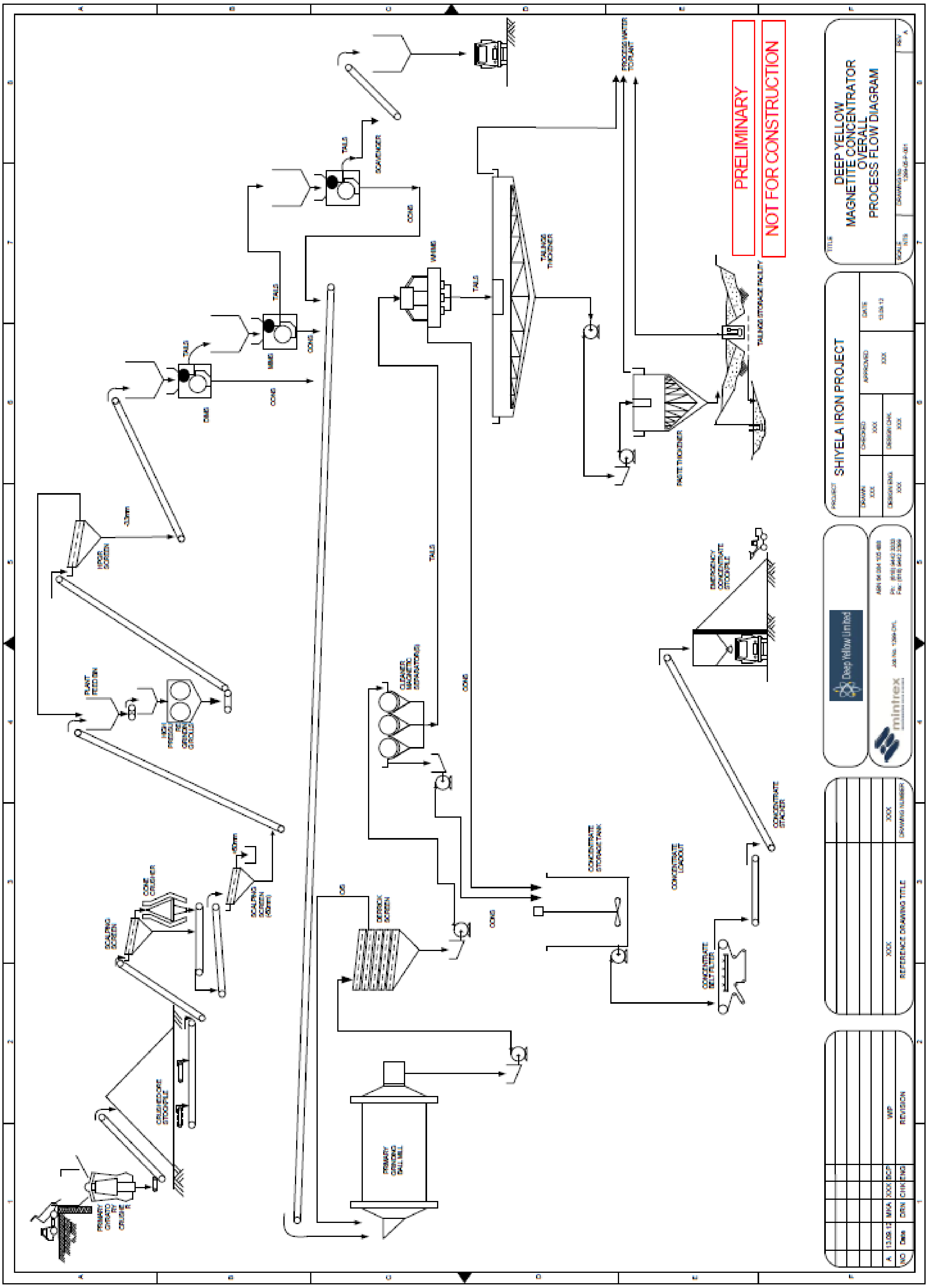

Figure 2: Shiyela Project – Schematic Plant Layout

Figure 3: Shiyela PQ Core Showing Hematite

Figure 4: Shiyela PQ Core Showing Folded Magnetite Bands

ENDS

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For further information on the Company and its projects - visit the website at www.deepyellow.com.au

About Deep Yellow Limited

Deep Yellow Limited (DYL) is an ASX-listed, advanced stage uranium exploration company with projects in the southern African nation of Namibia. It also has a listing on the Namibian Stock Exchange.

Deep Yellow's focus is in Namibia where its operations are conducted by its 100% owned subsidiary Reptile Uranium Namibia (Pty) Ltd (RUN). Its flagship is the Omahola Project where it is conducting resource and reconnaissance drilling along the high grade Ongolo– MS7 Alaskite trend. It is also evaluating a stand-alone project for its Tubas Sand deposit utilising physical beneficiation techniques it successfully tested in 2011.

In Australia the Company owns the Napperby Uranium Project and numerous exploration tenements in the Northern Territory and in the Mount Isa District in Queensland.

APPENDIX 1: PROCESS FLOW DESCRIPTION

Overview

The process plant circuit is designed to produce 2 Mtpa magnetite concentrate from an ROM feed of 7.52 Mtpa magnetite/hematite ore at a weight recovery of 26.6% including plant yield factors. The laboratory test work using the Davis Tube Recovery (DTR) indicated the final product will need a P80 of 250 micron to produce a moderate grade iron (>65.0% Fe) with an acceptable silica grade. The grade of final product will vary with the ratio of magnetite and hematite in the feed material with the hematite producing the low grade material.

Ore will be crushed to -200mm in an open circuit Gyratory crusher. The ore will be further crushed in a secondary cone crusher to -32mm (80% -25mm) followed by a High Pressure Grinding Roll (HPGR) circuit for further size reduction. The product will be screened at 3mm with the oversize returned to the HPGR for further size reduction.

The product will be concentrated using Dry LIMS and MIMs producing a dry tails which will pass through a scavenging MIMS before being conveyed to a bin for self-loading onto haulage trucks for disposal.

The combined preconcentrate is conveyed to the mill where it will be ground to 80% -250 micron in an overflow mill closed by Derrick screens. The under flow will be pumped to magnetic separation while the oversize will return to the mill. The cleaner magnetic separators will be produce a cleaned magnetite fraction and the tailings will be combined and feed to the GRZM Jigging WHIMS. The concentrate will be combined with the magnetite concentrate in an agitated storage tank ahead of the filter plant where two belt filters will produce a concentrate of 9% moisture for transfer to a truck loading bin.

Slurry tailings will be thickened in a conventional thickener and the thickened slurry pumped to the paste thickener situated near the tailings dam area. Water from both areas will be returned for recycling while the paste will be disposed using central thickened discharge pipework with little water expected to be recovered from the dam. The material may be combined with mine waste to provide a more effective disposal method.

Plant Design

The flowsheet shown in Figure 2 shows the process in a pictorial schematic. It comprises the following unit processes:

- Primary crushing;
- Secondary crushing;
- HPGR;
- Dry LIMs, MIMS, Scavenging MIMS
- Ball milling;
- CMS, WHIMS;
- Filtering of final mag cons;
- Tails dewatering.

Primary Crushing

The Run of Mine ore (ROM) is trucked to the ROM pad and deposited into the Primary Crusher and feeds a gyratory crusher (Metso Gyratory Crusher- 54/75), which will accept a maximum lump size of ~ 1m and give a product with P80 of 165mm. The tonnage capability of the primary crusher is based on operating at a peak capacity equivalent to achieving the target tonnage in 16 hours per day to cope with the variation in truck arrival times at the crusher. This is nominally 6000 hours per year. The crushed product will be conveyed to the crushed ore stockpile via an inclined fixed point stacker conveyor.

The stockpile will be 25 m high, with a live capacity of 10,400 tonnes or 8 hours of feed. The live storage has been designed to absorb delays caused by blasting interruptions, crib changes, shift changes etc and smooth out operations caused by the batch nature of the mining operation. Further, by dozing the remaining stockpile, the ore supply can be extended to allow for a 48 hour maintenance window shutdown.

Secondary Crushing

The crushed ore will be reclaimed from the stockpile via a single tunnel reclaim system. The system will consist of a conveyor with two extraction ports housing apron feeders. The advantages of having two extraction ports per conveyor are the reduction of the stockpile dead area, (the increase of the live area) and better utilization of equipment.

The ore to the secondary crusher is pre-screened to bypass the -32 mm material but otherwise the cone crusher operates in open circuit, and will reduce the stockpile product from 80% passing 165mm to 80% below 25 mm. The product will discharge to a scalping screen screening at 50mm – this prevents oversize created during occasional crusher disturbances reaching the HPGR circuit which can damage studs. The oversize material will be put to a conical stockpile for mechanical recovery.

The secondary crushed material will be monitored by a metal detector, which will be coupled to a high ramping tramp metal removal magnet. The device operates on the principle that as both metal and ore could be picked up, the ore, with a lower magnetic susceptibility, will be released back onto the feed conveyor as the device rotates out of the magnetic field,(because of the weaker magnetic field strength) while the metal is removed and drops into a skip.

The secondary crusher product will be fed into the High Pressure Grinding Rolls (HPGR) feed bin via conveyor.

HPGR and Primary Grinding

From the HPGR bin the feed is split into two streams to feed two moderate sized HPGRs (1.4 x 1.4m machines). The concept design is that after passing through the HPGR the ore is dry screened to recycle oversize through the machines again. Two screens will be used (3.5m x 7m – probably of Binder Bivitec design) to produce a -3.3mm product. A third bin will be provided at the screening station so that material can be recycled to the HPGR without screening to use the full capability of the machines.

Dry Magnetic Separation Circuit

The DMS section will recover both magnetite and hematite fractions and reject on average 30% of feed weight but up to 50% on occasions for a loss of less than 5% of the contained Fe.

Material will be elevated to the DMS building which contains 8/10 sets of dry LIMS/MIMS units – set up in double drum units. The concentrate streams are combined while the tailings will pass over a set of four scavenger MIMS to recover any misreporting material. The bins above the LIMS/MIMS units hold roughly one hours feed and the material is spread evenly over the rolls by using a rolls feeder.

The dry LIMS circuit are expected to be 1200gauss rolls while the MIMS rolls at 7,500gauss.

The final dry tailings will be transported by conveyor to a bin which can self load mine haulage truck for disposal on the waste dump or tailings dam area.

Ball Mill Grinding

The preconcentrate will be conveyed to the ball mill circuit where they will be ground to 80% below 250 micron. The circuit will be closed by Derrick screens, screening at roughly 300 micron with the oversize gravitating back to the mill, flushed by the mill water addition stream. The mill size is expected to be 8.55m long and 6.1m in diameter with a 5.6MW motor.

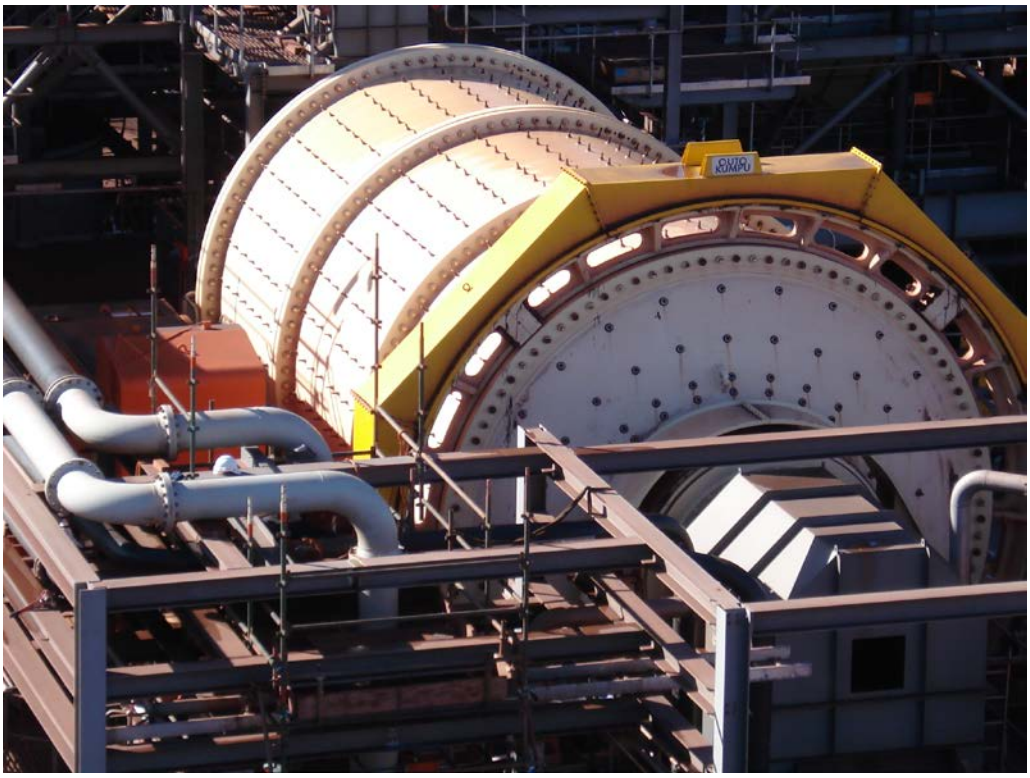


Figure 1: The ball mill at OneSteel in Whyalla

The product from the mill is diluted to 60% solids and screened with the undersize being transferred to the cleaner magnetic separator section.

The mill receives both magnetite and hematite concentrates and this stabilises the tonnage as the ore switches from hematite rich to magnetite rich.

Cleaner Magnetic Separation Circuit

In this circuit the magnetite is first removed and the tailings then treated to recover the hematite. Density control will be important to ensure that the tailings are not too dilute and overwhelm the GRZM WHIMS units.

Since the material feeding the cleaner magnetic separators (CMS) is relatively coarse then no slime removal is needed and hydroseparators have been removed from the original concept circuit. The CMS units are still triple drum units and the final drum produces a final product which is pumped to the filter feed storage tank.

There are 6 CMS units consisting of 1.2m diameter x 3m long triple drum units. The CMS separators will reject the non-magnetics generated by the grinding in the ball mill circuit to provide a clean magnetic product. The concentrate off the last drum will be pumped to the filter feed storage tank. The CMS non-magnetics will be transferred to the GRZM Whims.

Figure 2: A CMS drum in operation at OneSteel in Whyalla

The GRZM is a double pulsing aerating flow WHIMS which is designed to collect and clean the concentrates in one machine. SLON is more well known machine but only has the pulsing action and is hence less efficient. Testwork and microscope observations shows that the hematite is liberated and just needs good cleaning to reach grade. It is interesting that during the HL testwork phase that grades over 64% Fe were achieved in the hematite.

It is assumed that the GRZM will upgrade the Fe in the hematite to 61% Fe – based on the testwork with a yield of over 90%.

Concentrates will be pumped to the agitated storage tank ahead of the filters.

Concentrate Handling Circuit

The concentrate will be fed from the concentrate storage tank to the belt filter at 60% solids by weight where it will be filtered to a moisture content of 9%. The filter cake will then be conveyed into a 250t load out bin with excess overflowing against the bin wall to create an emergency stockpile.

Tails Circuit

The primary tails disposal system consists of:

- Dry tails from the DMS section;
- The tails thickener;
- The paste thickener.

The feed to the wet tails disposal system will consist of the non- magnetic fractions from the GRZM which will be roughly 66% of the total tails.

Flocculant will be added to the thickener and the fine solids will settle. The thickener underflow (60% solids by weight) will be combined with the tailings cyclone underflow and both will be pumped to the tailing treatment area at 55% solids by weight.

From the thickener the thickened tailings will be pumped to the paste thickener on the edge of the dam area where the slurry will be thickened to 75% solids. The paste thickener underflow slurry will be pumped to the Tails Storage Facility (TSF) by using positive displacement pumps.

The tailings will be discharged using a central thickened discharge concept pumping to one of two outlets to form a beach, at an angle of 3/4 degrees to the horizontal. The water run-off will run down the beach to a coffer dam situated at the base of the tailings facility. This style of tailings disposal will be relatively economical because it does not require a high water seal dam wall, but still provides a means of recovering residual water and any run off due to rain. Most of the process water will be recovered by the tailings thickener, with some water recovery by the paste thickeners and at the coffer dam wall.

APPENDIX 2: PROCESS FLOW DESCRIPTION

MINTREX MEMORANDUM ON METALLURGICAL YIELD AND RECOVERY

1. INTRODUCTION

Metallurgical testwork has been conducted over a series of programmes on the Shiyela deposit. This has encompassed testwork on the magnetite, recovery of hematite from magnetite tailings and recovery of hematite from a predominant hematite source.

The testwork has resulted in a plant design which can recover both magnetite and hematite – initially by dry magnetic separation at two fields strengths followed by a relatively coarse grind to 80% passing 250 micron followed by low intensity magnetic separation (LIMS) and low strength wet high intensity magnetic separation (WHIMS). The magnetite is recovered to a concentrate grade of 68% Fe and the hematite to a grade of 61% Fe. This has been demonstrated on a range of samples ranging from 5% Fe in feed and up to 40% Fe in feed and with varying proportions of magnetite and hematite.

From this an estimate of yield and product grade from the resource grade can be made.

2. BASIS OF RESULTS

2.1 Magnetite Testwork

The magnetite testwork covered samples from the magnetite zone and from samples in the hematite zone but which contained some magnetite

Six composites were provided to represent the magnetite zone and gave the following data:

80% Passing							
Micron	%Fe	%SiO_2	%Al_2O_3	%CaO	%MgO	%MnO	%P
250	67.32	3.28	1.62	0.18	0.59	0.76	0.025
150	69.53	1.22	1.26	0.07	0.48	0.78	0.009
75	70.18	0.73	0.91	0.06	0.41	0.78	0.004
45	70.32	0.81	0.77	0.03	0.35	0.78	0.005

At 250 micron, there was a low grade result and when this was excluded, the grade increased to 68.3% Fe and 2% SiO_2.

When the hematite samples were tested then the grade of the magnetite part averaged 68.6% Fe at 80% passing 250 micron. When the very low grade hematite samples were tested the magnetite part of these samples averaged 66.9% Fe which increased to 67.45% Fe when a very low sample of 63% Fe was excluded.

Since the grade is adjustable by slight changes in grind which is within the capability of the plant then 68% Fe has been adopted as the nominal grade of the magnetite after grinding to 80% passing 250 micron.

Since the plant has a 1200 gauss collection of magnetite followed by a 7000 gauss dry magnetic separation and 3000 gauss WHIMS in the wet section – it has been assumed that all magnetite measured by the Davis Tube (3000gauss) will be collected in the plant – i.e. 100% yield.

2.2 Hematite testwork

Hematite testwork has looked at hematite recovery from magnetite tailings and hematite recovery from ores classed as hematite.

2.2.1 Early work

In the magnetite tailings the testwork done at the time did not reach very high grade and would only be salable when mixed with magnetite. The Fe yield averaged about 72%

2.2.2 Hematite Sample – determining conditions

A programme was conducted to test the possible process route which involved:

Testing three sizes for DMS and MIMS dry magnetic separation
Testing the dry concentrate at four different sizes using:

- Heavy liquid
- Tabling
- Three different gauss levels

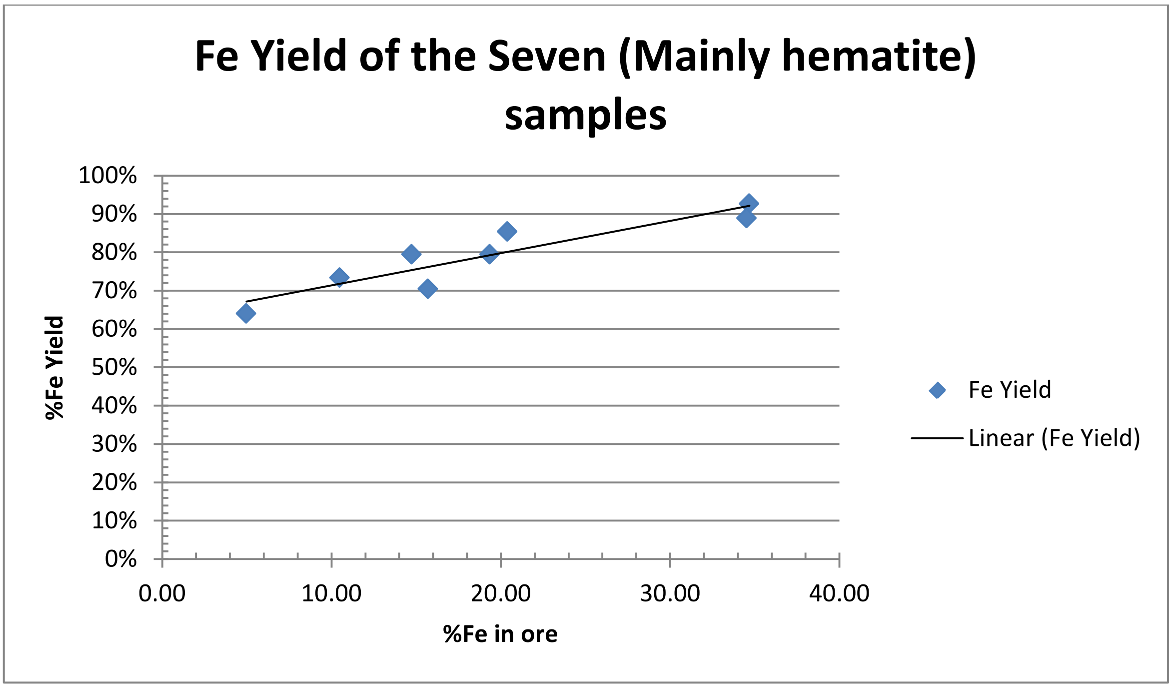
With the following results – the final process selection being a grind of 80% passing 250 micron and a WHIMS gauss of 3000gauss

Grade of Concentrate						
Passing Size	3,000 G	5,000 G	10,000 G	HL	Gravity	Gravity 60%
500	55.7	52	50.8	64.6	60.1	
250	**63.6**	**60.2**	**56.4**	**66.3**	**64.4**	**61.4**
125	62.4	61.8	57.4	65.6	63.6	
106	61.5	60.6	57.1	65.4	62.9	
Fe Yield to Concentrate						
Passing Size	3,000 G	5,000 G	10,000 G	HL	Gravity	Gravity 60%
500	96.8%	97.3%	98.2%	98%	59%	
250	**92.2%**	**94.2%**	**96.5%**	**95%**	**73%**	**86%**
125	89.5%	93.1%	96.1%	92%	94%	
106	82.7%	90.3%	94.2%	88%	84%	

From this data a conservative estimate was made of an 85% Fe yield and with 68% Fe from the magnetite and 61% Fe from the hematite. The hematite grade was 62.5% Fe at the target conditions and magnetite from the same sample was 68.6% Fe.

From this work seven low grade samples of different grades, mainly hematite samples were tested by the proposed plant process design.

The Fe yield to a grade above 61% Fe is shown in the graph below – essentially all samples exceed 70% Fe yield other than when the feed grade dropped below 5% Fe



The seven samples gave the following results – where the predicted result assumed 85% Fe yield of the feed Fe, with the DTR weight recovery assumed to be the magnetite proportion at 68% Fe and the remaining Fe being hematite at 61% Fe

Performance to Final product									
		A	B	C	D	E	F	G	Average
	%Fe in Feed	14.74	4.97	10.49	34.68	34.52	15.70	20.39	19.4%
Grade	Measure	62.2%	61.1%		63.5%	64.3%	65.2%	61.7%	63.0%
	Predict	*61.6%*	*61.1%*	*61.5%*	*68.0%*	*61.4%*	*63.0%*	*62.7%*	*62.8%*
Fe Yield	Measure	79.5%	64.0%	73.4%	92.7%	88.9%	70.5%	85.4%	79.2%
	Predict	*85%*	*85%*	*85%*	*85%*	*85%*	*85%*	*85%*	*85.0%*

The grade was slightly higher than predicted and the yield was slightly lower on average – though these samples were predominantly less than the mine average.

However the sample which had a grade closest to the mine grade of 20.7% Fe – sample G – met the 85% Fe yield predicted even with low levels of magnetite.

3. ESTIMATE OF METALLURGICAL RECOVERY ACROSS THE RESOURCE

The proposed procedure is:

- 85% of the Fe will recovered to concentrate
- The Davis Tube concentrate weight recovery is assumed to produce concentrate at 68% Fe. (The DTR is conducted at 80% -45micron so is closer to 70% Fe)
- The remaining Fe is estimated at 61% Fe to give a weight yield from the hematite
- The two concentrates are mixed to give a total weight recovery and grade.
- If no DTR is available then assume it is all hematite

The alternative method is to assume 100% Fe yield from the magnetite and 70% from the hematite – more cumbersome process – though the two are compared below

			85% Yield Method	
A	Head Grade		20.3	%Fe
B	DTR Wt Recovery		14.2	% to Cons
D	Fe units available (Head x 0.85)	A x 0.85	17.26	
E	Fe units with Mags	B X 0.68	9.66	
F	Fe units for Hematite	D - E	7.60	
G	Wt at 61%	F/0.61	12.5	
H	**Total Weight Recovery**	B + G	**26.7**	
	Grade	D/H	**64.7%**	

			100% and 70% Yield	
I	Magnetics	B *0.68 *1.0	9.7	
J	Remaining units	A - I	10.6	
K	Fe units Yield at Hematite	J * 0.7	7.5	
L	Wt at 61%	F/0.61	12.2	
M	**Total Weight Recovery**	B + L	**26.4**	
	Grade	(B*0.68+L*0.61)/M	**64.8%**	

At the scoping level study the 85% method is probably the simpler to use. Both give a similar result and both are probably conservative. The following table shows that the Fe units recovered by the LIMS units exceeds the units predicted by DTR - for each of the samples – partly due to the coarser grind in the plant than for the DTR.

	A	B	C	D	E	F	G
DTR Fe Units	1.18		0.68	31.04	1.89	4.11	4.72
Fe units Recovered	1.68	0.02	1.01	34.59	3.39	6.67	9.16

Yours faithfully,
MINTREX

B C Povey
Principal Consulting Metallurgist

APPENDIX 3: SUMMARY SHIYELA JORC MINERAL RESOURCE ESTIMATE – DECEMBER 2012

Deposit	Category	Cut-off Grade	Tonnes (M)	Fe (%)	DTR (%)
M62 – Magnetite	**Indicated**	**10 wt% DTR**	**35.2**	**-**	**17.62**
	Inferred	**10 wt% DTR**	**9.4**	**-**	**15.75**
	Total		**44.7**	**17.33**	**16.37**
M62R – Magnetite	**Inferred**	**10 wt% DTR**	**9.3**	**16.30**	**17.40**
	Total		**9.3**	**16.30**	**17.40**
M63 – Magnetite	**Indicated**	**10% Fe**	**5.3**	**22.32**	**15.78**
	Inferred	**10% Fe**	**29.2**	**20.80**	**15.21**
	Total		**34.5**	**-**	**15.30**
M63 – Hematite	**Inferred**	**10% Fe**	**26.7**	**22.29**	**-**
	Total		**26.7**	**22.29**	**-**

Notes: Figures have been rounded and totals may reflect small rounding errors
Resources were reported using a 10% DTR wt% cut-off grade.
The DTR estimates are based on samples prepared at a grind size of 80% passing 45 micron.
Fe% - head assay of composited drill samples

Compliance Statement:

The information in this report that relates to Mineral Resources is based on information compiled by James Farrell who is a full-time employee of Golder Associates Pty Ltd and a Member and Chartered Professional of the Australasian Institute of Mining and Metallurgy. James Farrell has sufficient experience to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking, to qualify as a Competent Person as defined in the JORC Code (2004). James Farrell has relied on exploration data compiled by Dr Leon Pretorius who is the Managing Director of Reptile Uranium Namibia (Pty) Ltd and a Fellow of the Australasian Institute of Mining and Metallurgy. Dr Pretorius has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the JORC Code (2004). James Farrell has also relied on interpretation of metallurgical testwork compiled by Brian Povey who is a full-time employee of Mintrex Pty Ltd and a Fellow of the Australasian Institute of Mining and Metallurgy. Brian Povey has sufficient experience to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking, to qualify as a Competent Person as defined in the JORC Code (2004). James Farrell, Leon Pretorius and Brian Povey consent to the inclusion of this information in the form and context in which it appears.